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The Active Roof Supporting Technique of a Double-Layer Flexible and Thick Anchorage for Deep Withdrawal Roadway under Strong Mining Disturbance

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Abstract: Due to the strong disturbance of a mining face, the surrounding rock of the withdrawal roadway is susceptible to deformation and failure, which restricts the safe and efficient evacuation of mining equipment. To resolve this longstanding technical problem in mine production, an engineering investigation, numerical simulation, theoretical analysis, and other research methodologies were conducted in this study. Furthermore, the influence mechanism of mining-induced stress on the withdrawal roadway was revealed, the anti-disturbance principles of thick-layer anchorage of roadway roofs were elucidated, and a novel double-layer flexible support technique was proposed. The front abutment pressure, stress superposition, damage accumulation of the surrounding rock, and the fluctuation of mining-induced stress are the primary factors contributing to the significant deformation of the surrounding rock in a withdrawal roadway. However, the fluctuation of mining-induced stress has usually been ignored in previous studies, and it may be the most crucial cause of the significant deformation and instability of the surrounding rock. The thickness of the anchored rock beam is the most vital factor affecting the maximum subsidence and maximum tensile stress of the roof, and increasing the thickness of the anchored rock beam can significantly improve the stability and anti-disturbance performance of the roof. In the proposed double-layer flexible supporting technique, flexible steel strands serve as the carrier, which overcomes the constraint of the roadway height on the length of roof support components. The first layer of flexible support is used to construct a thick fundamental anchorage layer, while the second layer is employed to construct a thicker reinforced anchorage layer, facilitating the effective resistance of the roof against strong mining disturbances. The effectiveness of this technique was further validated through the application of an engineering practice. The research results have reference value for solving the difficult problem of mining roadway support.

Keywords: withdrawn roadway; strong mining disturbance; stress fluctuation; double-layer flexible support; thick anchor

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1. Introduction

The withdrawal roadway is a specialized mining roadway arranged at the stop-mining line of a coal mining face. It serves as a direct connection to the gob after the completion of mining operations, providing crucial spatial support for withdrawing the mining equipment [1,2]. The mining areas in the Inner Mongolia Autonomous Region and Shaanxi Province are the core regions of coal production in China, and the deep mining method has been widely employed in these mining areas [3–5]. To ensure the safe and efficient mining of deep coal, the mining equipment should be transported and withdrawn efficiently. The roof control of the withdrawal roadway is an important guarantee for the safe and rapid retreat of the mining equipment from the working face. However, during the final mining period, the withdrawal roadway will be connected to the coal mining face, which will cause severe deformation, fracture, and instability of the surrounding rock in this roadway due to strong mining influence. This poses a formidable challenge for roof control and has been a persistent issue affecting mine production [6–8].

The weighting characteristics of the withdrawal roadway mainly depend on the distribution of the front abutment pressure in the working face [9,10]. The classic mine pressure theory points out that the front abutment pressure along the advancing direction of the working face can be divided into the following three zones: the stress reduction zone, the abutment pressure zone and the original-rock stress zone. Researchers have conducted many studies on front abutment pressure. It has been concluded that the value and distribution range of the stress reduction zone and abutment pressure zone significantly affect the stability of the surrounding rock of the withdrawal roadway. Yang et al. [11] studied the influence of the fracture location of the main key layer of the roof on the stability of the surrounding rock of the withdrawal roadway and proposed a differentiated support plan for the withdrawal roadway. He et al. [12] pointed out that the stop-mining line should be reasonably designed to ensure that the main key layer fracture line is located behind the hydraulic support. Li et al. [13] put forward the treatment method for the manifestation of strong strata behaviors by using hydraulic fracturing technology to break the key stratum and to protect the reserved roadways. Under the mining conditions of a 15-m-thick coal seam in the Datong mining area, Xie et al. [14] conducted a case study on the distance between the stop-mining line and the location of the main key layer fracture line of the roof. It was concluded that, when the stop-mining line distance is slightly greater than the length of the key block, the stability of the surrounding rock in the withdrawal roadway is optimal. The above studies all posit the idea that the stress distribution of the surrounding rock in small roadway areas is related to the fracture morphology of the roof key layer.

In the aspect of support structure optimization, the combination of anchor bolts and cables has been employed to control the deformation of the surrounding rock of the roadway under strong mining disturbance. After installation, the anchor bolts and cables provide supporting resistance to the surrounding rock. With the deformation of the surrounding rock, the support resistance is increased. Thus, the utilization of anchor bolts and cables can improve the strength of the surrounding rock timely and effectively, prevent the early separation of the surrounding rock, and control the deformation of the surrounding rock. Wang et al. [15] installed anchor cables within deep and stable rock layers to stabilize a larger range of rock masses in order to suppress deformation, which fully utilized the bearing capacity of the surrounding rock itself. These studies suggest that the purpose of bolt–cable support is to give full play to the bearing capacity of the rock mass.

Based on the typical deep coal seam conditions in a large-scale mine of 10 million tons in western China, the surrounding rock control of the withdrawal roadway under strong mining disturbance was investigated through theoretical and technological research in this study. The mechanism of large deformation in the withdrawal roadway that occurred during mining was analyzed, and a novel active roof control technique of a double-layer flexible and thick anchorage was proposed for the deep withdrawal roadway under the strong mining disturbance.

2. Research Background

2.1. Roadway Layout and Geological Conditions of Test Points

The test point was located in working face 21405 of the Hulusu coal mine. This working face was the first mining face in the fourth panel of coal seam 2-1 of the mine. Figure 1 shows the roadway layout of working face 21405. The daily progress of the working face was 5.6 to 9.6 m. The two entries were abandoned behind the longwall progress. The withdrawal roadway was arranged near the stop-mining line for the withdrawal of the production equipment after the completion of mining. A protective coal pillar with a width of 360 m was left between the withdrawal roadway of working face 21405 and the main roadway in the panel area. The value of the pillar width was greater than that of the conventional value (half the width of the working face), which facilitated the final layout of a longitudinal longwall face after all of the working faces to the right were completed.

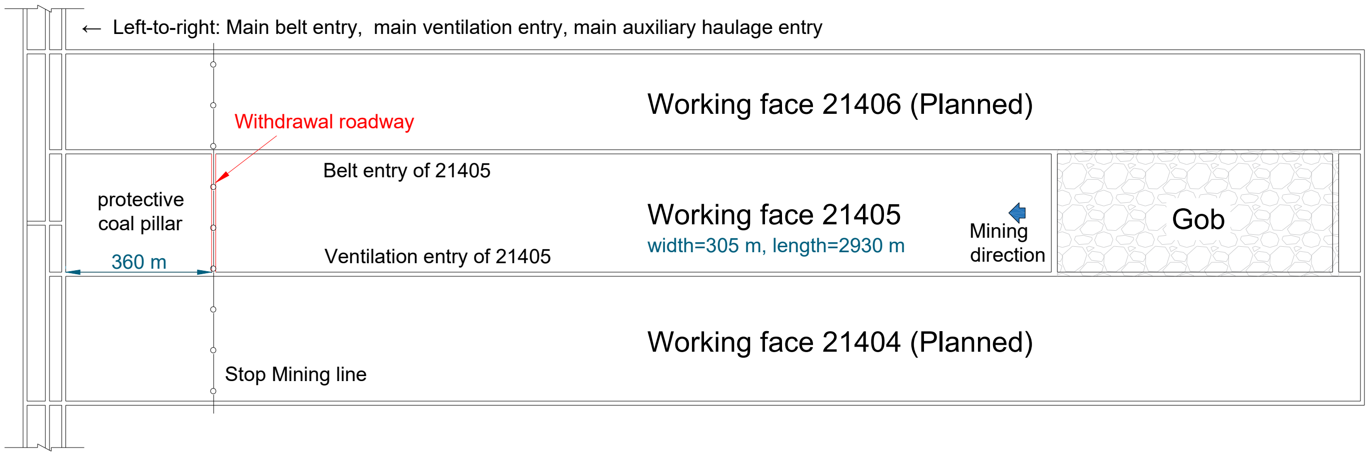


Figure 1. Roadway layout and mining environment.

The buried depth of the working face was 650 m, and the thickness of the coal in the final mining area was 3.4 m. The cross-sectional shape of the withdrawal roadway was rectangular, with a width of 4.8 m and a height of 3.4 m. It was driven using a roadheader for high efficiency construction. In the withdrawal roadway, the roof of the coal seam was composed of medium sandstone, fine sandstone, and coarse sandstone, and the floor was composed of mudstone or sandstone. Figure 2 shows the occurrence situation of the rock strata, and Table 1 shows the mechanical parameters of the rock layer. As depicted in Figure 2, the withdrawal roadway will experience the most significant mining impact during the final mining period, that is, the working face will be continuously advanced until it connects with the withdrawal roadway. During this period, the front abutment pressure of the working face will gradually approach the withdrawal roadway, and the impact on the stability of the surrounding rock in the withdrawal roadway will become increasingly significant.

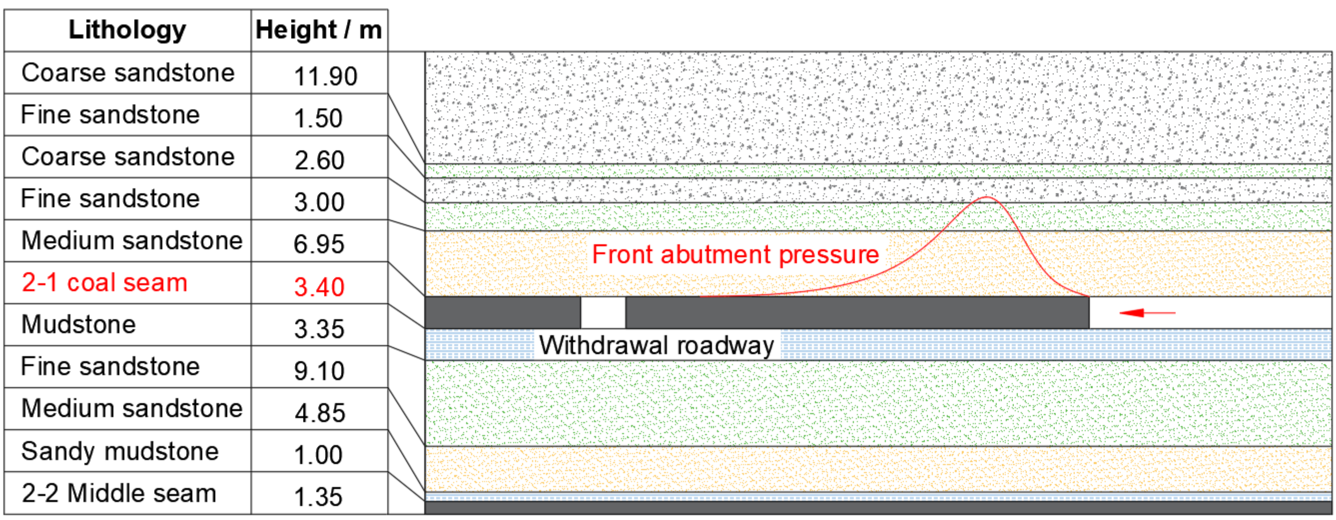


Figure 2. Profile view of the rock strata and the relationship between the withdrawal roadway and the working face.

Table 1. Mechanical parameters of rock strata.

Lithology	Density ($kg \cdot m^{-3}$)	Bulk Modulus (GPa)	Shear Modulus (GPa)	Tensile Strength (MPa)	Cohesion (MPa)	Internal Friction Angle (°)
Sandstone	2550	10.35	7.74	2.4	3.15	36
Coal seam	1400	3.05	2.13	1.0	0.82	26
Mudstone	2500	5.94	4.05	1.8	1.53	32

In order to effectively control the stability of the surrounding rock in the withdrawal roadway, the following three key tasks need to be accomplished: (1) understanding the causes behind the strong rock pressure and the significant deformation of the surrounding rock in the withdrawal roadway; (2) selecting a reasonable construction time to ensure that the excavation of the withdrawal roadway is not affected by the mining; and (3) implementing a scientific support plan to control the deformation of the surrounding rock of the withdrawal roadway during the mining activities.

2.2. Previous Support Techniques and Their Defects

Previously, complex supporting methods were used to cope with the influence of strong mining on the withdrawal roadway in this mine. During the roadway excavation, anchor bolts and cables were mainly used to complete the first support. Before mining, the anchor cables were used for the second reinforced support. During mining, the single hydraulic support columns, stacked supports, or concrete pier columns were used for the third reinforced support.

However, this composite support technique exhibits the following limitations: (1) In combined bolt–cable support, the mechanical properties of the bolts and the cables differ greatly, leading to synergistic difficulty during the bearing period; moreover, a limited number of anchor cables may experience breakage, due to excessive load bearing. (2) The completion of three supports requires the successive organization of three teams, resulting in the significant allocation of the construction resources and an extended construction period. (3) The utilization of a single hydraulic prop, stacked support, or a concrete pier column occupies the space of the withdrawal roadway, thereby reducing the efficiency of the equipment withdrawal. (4) The composite support method has high labor intensity and a high cost. (5) The composite support method has not completely solved the problem of the large deformation of the surrounding rock in the withdrawal roadway under strong mining disturbance, and the local deformation and failure of the rock mass increase the difficulty of repair and support reinforcement. (6) The maximum deformation of the roof and floor exceeds 1000 mm, and the maximum deformation of the two ribs exceeds 800 mm. The deformations are accompanied by fractures that loosen the surrounding rock and make the structure unstable. In this case, there is a safety risk of the roof falling in. The maintenance and control of the roof stability in withdrawal roadways under the influence of strong mining is still challenging, and the existing supporting method is intricate, necessitating an urgent resolution for the roof support of the withdrawal roadways.

3. Impact of Mining on Withdrawal Roadway

3.1. Modeling of the Spatial Structure of the Withdrawal Roadway and the Stope

A FLAC numerical simulation model was established based on the engineering conditions of the withdrawal roadway, as shown in Figure 3. The upper boundary of the model was subjected to stress constraints, with a vertical stress of 14 MPa, and all of the other boundaries were subjected to displacement constraints. The rock strength criterion used was the Mohr–Coulomb criterion. During the simulation process, the working face gradually advanced, with a cumulative advancement of 300 m, until it was connected with the withdrawal roadway. During this period, the changes in the stress and deformation of the surrounding rock were monitored for the subsequent theoretical analysis.

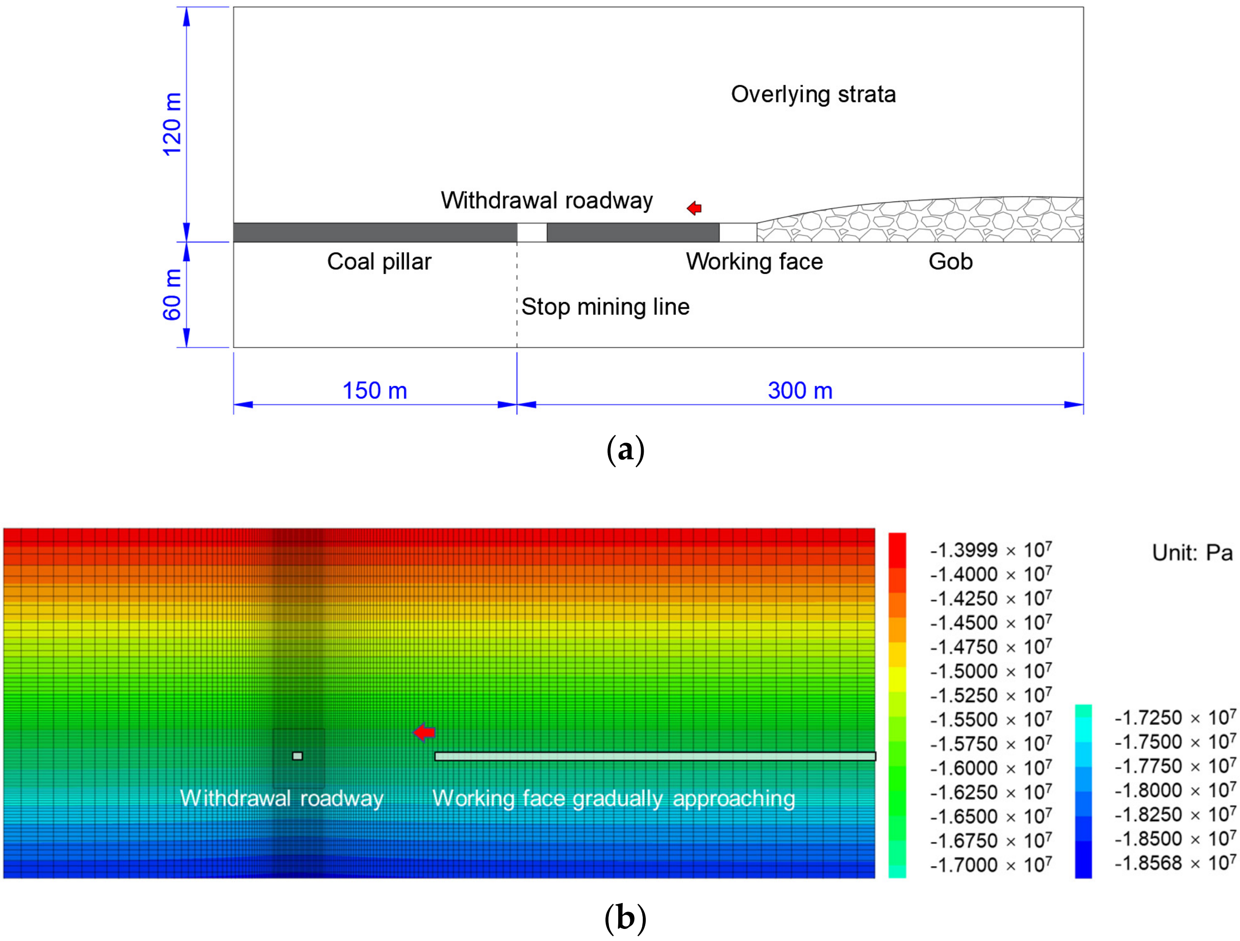


Figure 3. Numerical calculation model of the withdrawal roadway and the working face: (**a**) The spatial relationship and size of the model; (**b**) Numerical modeling and in situ stress field.

Combined with the numerical simulation, this study focuses on the following three aspects: (1) the influence mechanism of the mining-induced stress on the strong deformation of the surrounding rock in the withdrawal roadway is analyzed in order to deepen the understanding of the problem of strong mining; (2) the influence range of the advanced mining on the working face is obtained in order to determine the reasonable excavation timing for the withdrawal roadway; and (3) the evolution law of stress and deformation in the surrounding rock of the withdrawal roadway during critical mining operations is summarized in order to determine the supporting parameters.

3.2. Influence Mechanism of Mining-Induced Stress on the Withdrawal Roadway

This study has demonstrated that there are four primary factors contributing to the severe deformation of the surrounding rock in the withdrawal roadway during mining activities, as follows:

(1) Abutment pressure. After roadway excavation, the abutment pressure is formed on both sides of the coal bodies. After coal seam mining, the roof of the coal seam undergoes periodic fracture, leading to stress transfer from the gob to the periphery. Among them, there is a high value of front abutment pressure distributed in front of the working face. As shown in Figure 4, the peak value of the abutment pressure on both sides of the roadway is 22.8 MPa, which is 1.22 times higher than that of the original rock stress. The front abutment pressure significantly increases, and its peak value reaches 55.18 MPa, which is 3.25 times higher than that of the original rock stress. The abutment pressure caused by coal mining is 2.4 times higher than that caused by excavation. Therefore, the withdrawal

roadway is more prone to deformation when it is disturbed by the mining of the working face. This is consistent with previous studies [16].

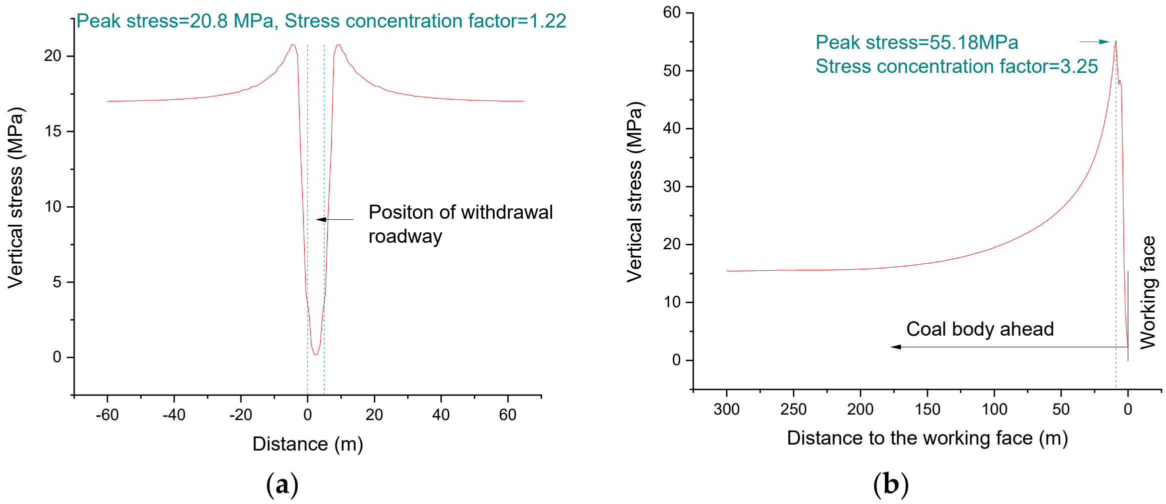


Figure 4. Distribution curve of abutment pressure: (**a**) The abutment pressure of coal mass on both sides of the withdrawal roadway; (**b**) Front abutment pressure of working face 21405.

(2) Stress superposition. When the coal mining face is still at a certain distance from the withdrawal roadway, the front abutment pressure has already been superimposed with the abutment pressure inside of the surrounding rock of the withdrawal roadway. Figure 5 shows the vertical stress evolution of the coal seam roof when the working face is gradually advanced to the withdrawal roadway. When the distance between the working face and the withdrawal roadway is 245 m, the stress variation in the withdrawal roadway is the smallest. When the distance is 180 m, the stress variation is relatively small. When the distance is 115 m, the abutment pressure on both sides of the withdrawal roadway increases slightly. When the distance is 55 m, the abutment pressures on both sides are superimposed onto each other, and these values show a significant increase. When the distance between the two is 5 m, the position of the peak abutment pressure has already crossed the withdrawal roadway and reached the rear of the withdrawal roadway, and the peak stress is as high as 68.45 MPa. When the superimposed stress exceeds the bearing capacity of the surrounding rock (especially the coal body), a strong deformation of the withdrawal roadway occurs.

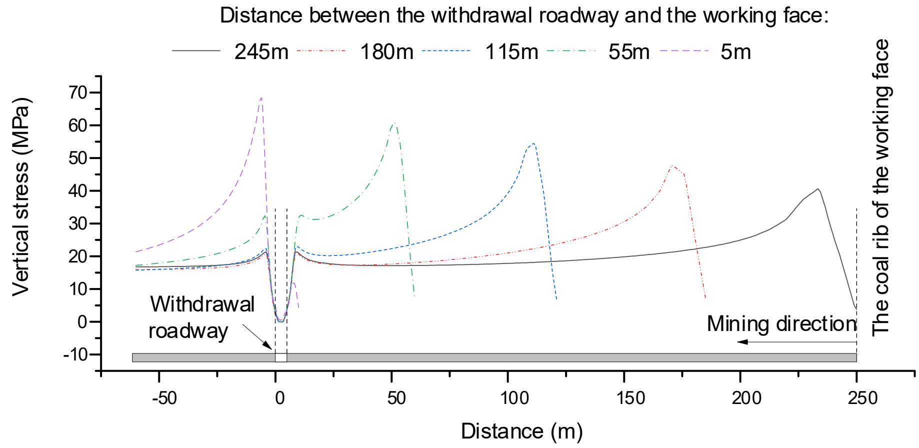
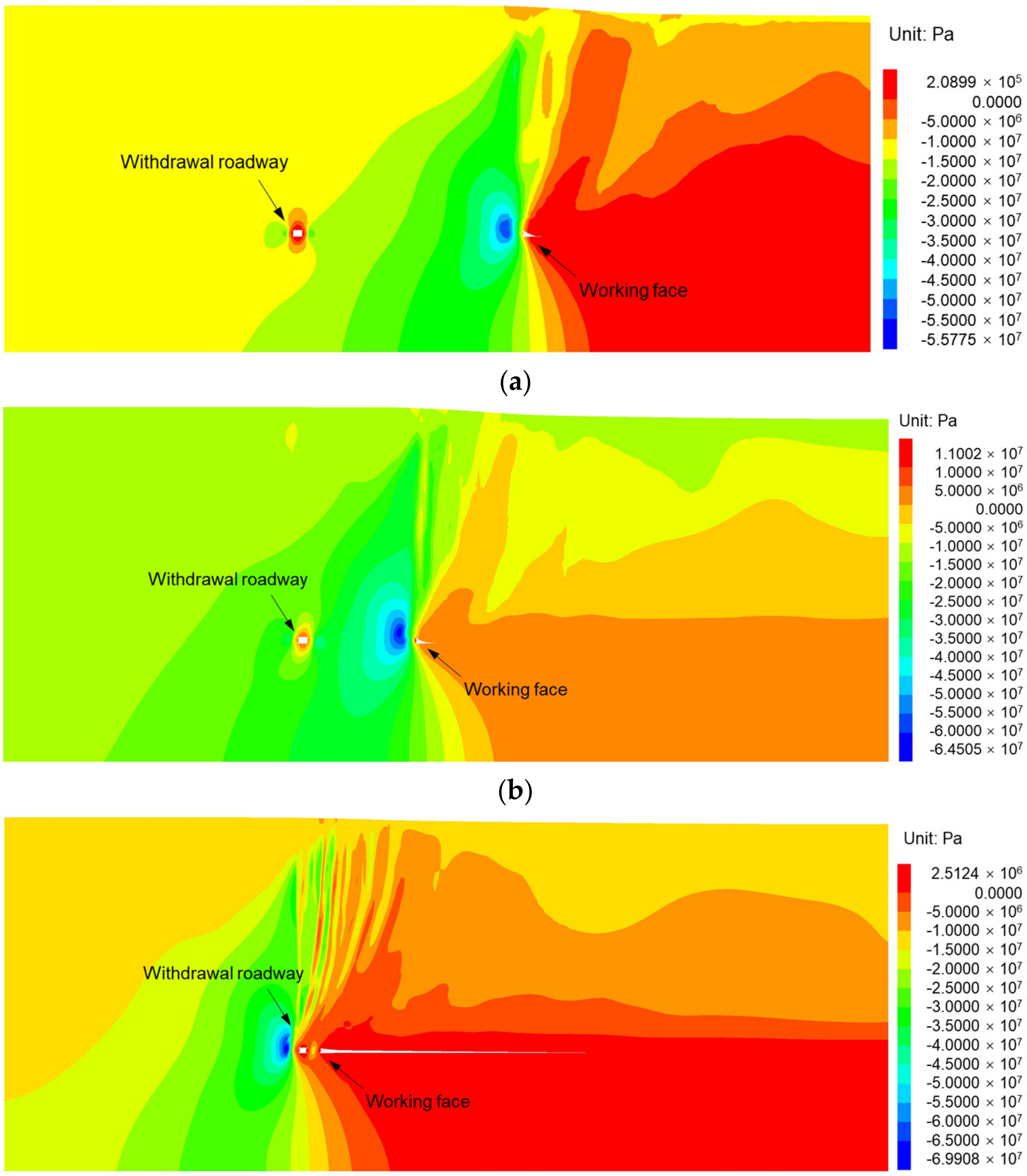


Figure 5. Vertical stress distribution curve at different distances between the working face and the withdrawal roadway.

(3) Damage accumulation. During the disturbance period, the surrounding rock of the withdrawal roadway exhibits stress-induced responses. The deformation and failure of the rock mass are the most evident manifestations. Due to the continuous occurrence of stress variations, rock mass damage continues to accumulate, ultimately resulting in large deformation and instability.

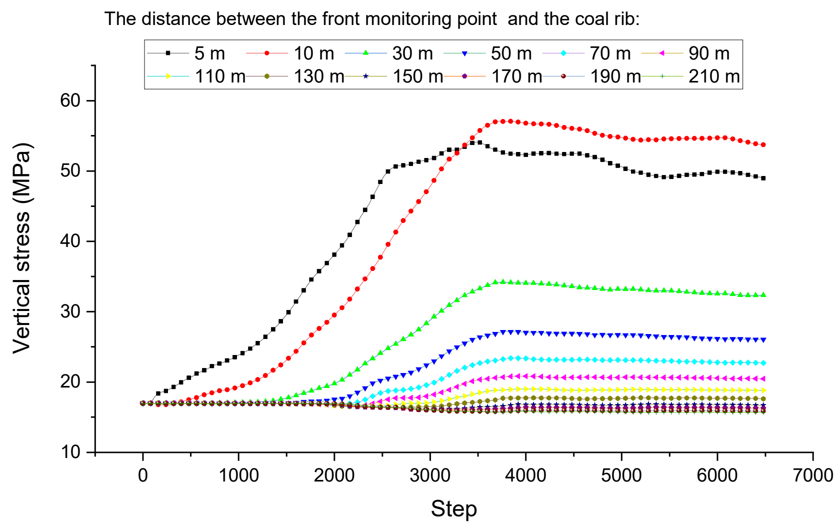
(4) Stress fluctuation. Although stress fluctuation has often been overlooked in previous research, it plays a crucial role in the deformation and instability of the roadway surrounding rock under strong mining disturbance. Figure 6 shows the stress distribution nephogram of the rock layer when the working face is advanced to a distance of 115 m, 55 m, and 5 m from the withdrawal roadway, respectivley. Figure 5 shows the stress value of the coal seam roof. At this time, the stress of each point in the figure seems to be determined, while the stress at every point in the rock mass undergoes a complex wave-like evolution process.



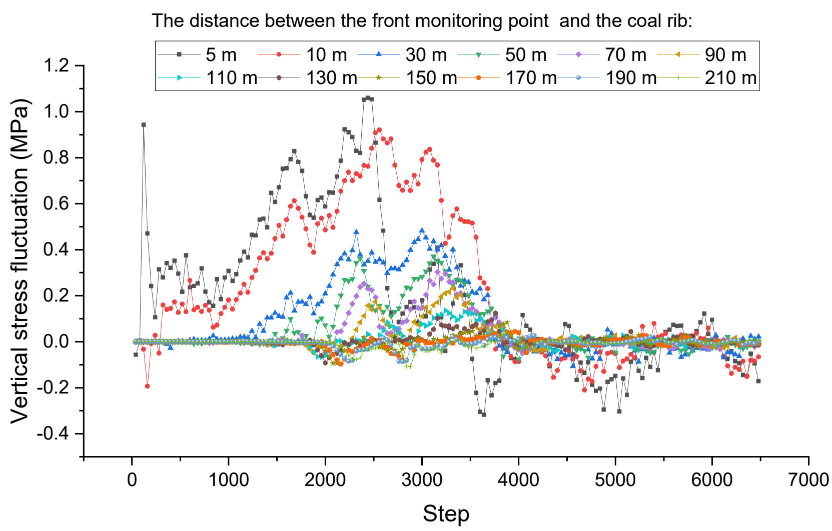
(c)

Figure 6. Vertical stress nephogram when the distance between the working face and the withdrawal roadway is (**a**) 115 m; (**b**) 55 m; (**c**) 5 m.

Figure 7a–c shows the stress evolution curve, stress fluctuation curve, and stress amplitude curve of the coal body at different positions in front of the mining work under different mining disturbances. In the area 130 m away from the front of the working face, the stress fluctuation of the coal body is very small, and the amplitude of stress variation is also small, with a maximum change value of 0.83–1.36 MPa. This area can be considered as a weak disturbance area. In the range of 130–30 m in front of the working face, the stress fluctuation of the coal body is significantly enhanced, and the amplitude of stress variation also increases rapidly, with a maximum change value of 2.43–17.25 MPa. This area can be considered as a moderate disturbance area. In the range of 30 m to 0 m in front of the working face, the stress fluctuation of the coal body is the strongest. The range of stress variation in this area is 37.19–40.34 MPa, which has exceeded the original rock stress by two times. Frequent stress fluctuations are equivalent to the frequent loading and unloading of coal and rock masses, which are more likely to induce cracks and deformation. This may be the most important cause of the deformation and instability of the roadways under strong mining disturbance.

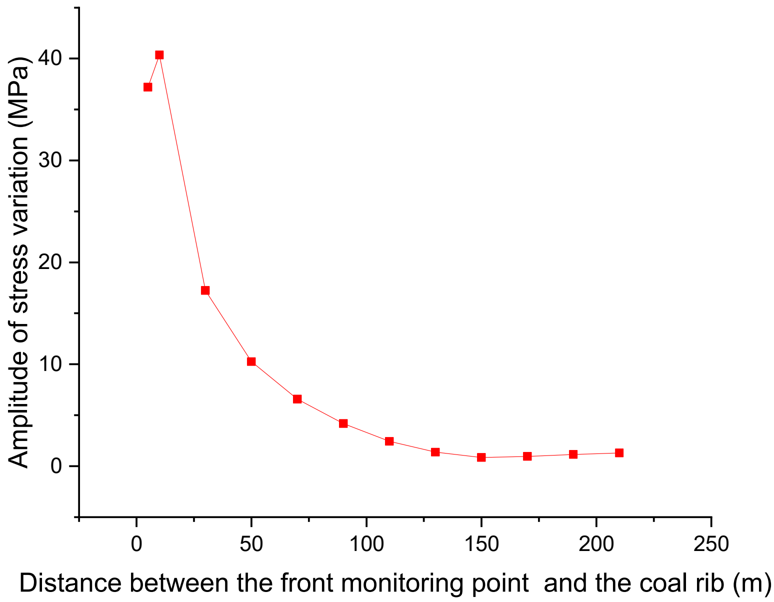


(**a**)



(**b**)

Figure 7. *Cont.*



(c)

Figure 7. The evolution of vertical stress during mining: (**a**) The evolution curves of vertical stress at different measuring points in front of the working face; (**b**) The evolution curves of stress fluctuation value; (**c**) The amplitude of stress variation.

3.3. Excavation Timing of Withdrawal Roadway

The majority of mines aim to excavate the withdrawal roadway later in order to concentrate the construction efforts on the preparation and extraction of the working face. To mitigate the impact of the mining activities on the working face, the determination of the latest feasible time for excavating the withdrawal roadway has emerged as a significantly crucial concern in this scenario. Based on the law of rock pressure and the numerical simulation results, the following can be elucidated:

(1) The influence range of the front abutment pressure on the working face should not exceed 1.0–1.2 times that of the width of the working face [17,18].

(2) From the analysis of the abutment pressure characteristics, the influence range of the front abutment pressure on working face 21405 is 142.7 m (0.47 times more than that of the width of the working face), while the influence range of the abutment pressure on the withdrawal roadway is 28.2 m (5.9 times more than that of the width of the withdrawal roadway).

(3) The stress superposition occurs when the distance between the working face and the withdrawal roadway is 115 m.

(4) At 130 m in front of the working face, the stress fluctuation of the coal body significantly increases.

From the perspective of facilitating the maintenance and control of the withdrawal roadway, the maximum distance of 171 m (which is the sum of the influence ranges of the abutment pressure between the working face and the roadway) can be selected as a safe distance to avoid significant disturbances. In other words, reasonable timing should be selected to ensure that the distance between the coal face and the withdrawal roadway is greater than 171 m when the excavation of the withdrawal roadway is completed. Considering that the excavation speed v_1 and the mining speed v_2 are not consistent, the following function can be used to determine the excavation timing:

$$L = \frac{305 v_2}{v_1} + 171 \quad (1)$$

When the distance between the coal mining face and the designated position of the withdrawal roadway reaches L, the excavation of the withdrawal roadway must commence. During on-site implementation, if there are changes in mining production (for example, if the excavation speed of the withdrawal roadway is slowed down or the mining speed of the working face is accelerated), this distance can be appropriately enlarged.

4. Double-Layer Flexible Support Technique and its Key Parameters

4.1. Anti-Disturbance Principle of Thick-Layer Anchorage

The front abutment pressure is induced by coal seam mining. When the distance between the working face and the withdrawal roadway is sufficiently small, the front abutment pressure overlaps with the abutment pressure of the withdrawal roadway, resulting in a further increase in peak stress [19–21]. When any point in the rock mass evolves into a new stress equilibrium state, it will produce frequent stress fluctuations, leading to cumulative damage to the rock mass. It can be seen that the reasons for the large deformation and failure of the withdrawal roadway caused by mining disturbance are complex. At this point, it is necessary to build a stable anti-disturbance support structure.

The coal seam roof exhibits typical layered characteristics, and, even with the same lithology, it is often subdivided into multiple rock layers due to the presence of weak interlayers within it. The individual layers are comparatively thinner, exhibit low stability, and are susceptible to deformation. By using an anchor bolt support, multiple thinner rock layers can transform into thicker and more integrated anchorage layers, and the stability and disturbance resistance of the roof can be greatly improved.

The roof rock layer can be assumed to be a simply supported beam to reflect the actual stress situation of the layered roof. Supposing that the thickness and the width of the anchored rock beam on the roof are h (m) and b (m), respectivley, I is the moment of inertia of the cross-section to the neutral axis (for a rectangular section $I = bh^3/12$, m^4), the load borne is q (MPa), the elastic modulus is E (MPa), and the roadway width is w (m), then the maximum roof subsidence $s_{\max}$ can be obtained as follows:

$$s_{\max} = 5qw^4/384EI = 5qw^4/32Ebh^3 \tag{2}$$

During the roof subsidence, the maximum bending moment is $M_{\max} = qw^2/8$ and the bending interface coefficient is $W = bh^2/6$. Therefore, the maximum tensile stress borne by the anchored rock beam of the roof $\sigma_{\max}$ is as follows:

$$\sigma_{\max} = M_{\max}/W = 3qw^2/4bh^2 \tag{3}$$

As shown in Equations (2) and (3), the thickness h of the anchored rock beam is a sensitive factor affecting the maximum subsidence and maximum tensile stress of the roof. If typical values are taken for the other parameters in the equation (q = 1.2 MPa, w = 4.8 m, E = 220 MPa, b = 1 m), then h has a significant impact on $s_{\max}$ and $\sigma_{\max}$, as presented in Figure 8. It is obvious that the larger the h, the smaller the maximum settlement and maximum tensile stress of the roof. Therefore, increasing the thickness h of the anchored rock beam is crucial for ensuring the safety of the roadway roof. The thicker anchorage not only mitigates roof deformation, but also alleviates the tensile stress by enhancing the overall bearing capacity of the roof.

Front abutment pressure can lead to rock failure in some cases. The important attribute of thick layer anchoring at this stage is that it can maintain the stability of the rock structure surrounding the roadway and provide support for further strengthening measures, such as grouting.

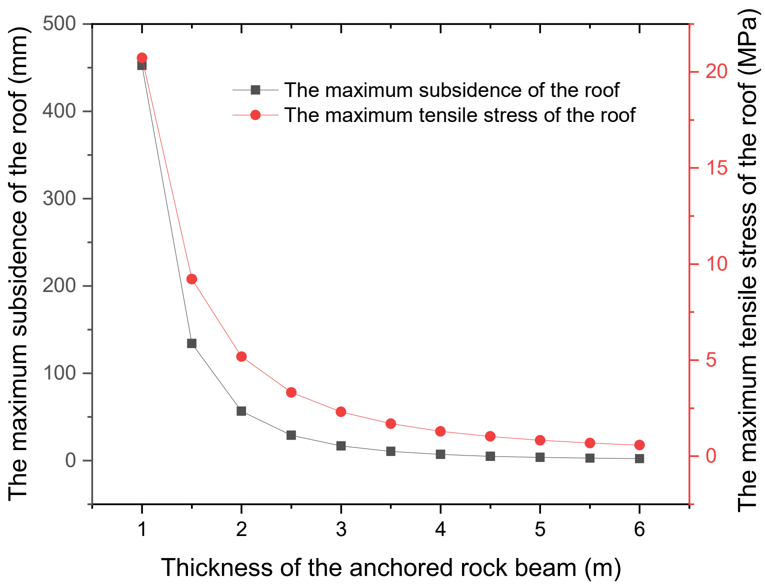


Figure 8. The relationship between the thickness of the anchored rock beam and the maximum subsidence/the maximum tensile stress of the roof.

4.2. Double-Layer Flexible Support Technique

Currently, a rigid anchor bolt made from non-longitudinal reinforced threaded steel has been extensively used in most roadway supports [22,23]. The ultimate length of this type of anchor bolt is constrained by the following three factors: (1) the anchor bolt's inability to bend; (2) the design height of the roadway limits the roof bolt length; and (3) a portion of the gangue in the excavation face is not transported out in a timely manner, but instead serves as a working platform for roof support (otherwise, the workers are unable to access the roof if the roadway elevation is too high). Consequently, the length of the roof anchor bolt is limited. Evidently, the support length of the rigid anchor rod is limited and fails to meet the demands of thick-layer anchorage.

The steel strands possess flexible properties and are capable of being bent. Therefore, the length of the steel strand is not limited by the height of the roadway or the available working space. The tensile strength of the steel strand is much higher than that of the threaded steel anchor bolt. Table 2 presents the mechanical parameters of the most commonly used anchor bolts and steel strands in coal mine roadways. The tensile strength and yield strength of the steel strands are 2.78 times and 3.2 times that of the rebar, respectively; therefore, the steel strands have a better bearing capacity. The cable support is adapted to variable load conditions [24,25], especially shear stresses, which may occur during mining disturbances. The utilization of steel strands for supporting the roadway roof enables not only effective thick anchorage, but also the application of high pre-tightening force. This approach significantly reduces the likelihood of bolt detachment, thus achieving robust anti-disturbance support.

Table 2. Mechanical parameters of bolt and cable most commonly used in coal mine roadway.

Materials	Tensile Strength/MPa	Yield Strength/MPa	Elastic Modulus/GPa	Elongation Rate/%
Bolts	670	500	210	20.0
Anchor cables	1860	1600	202	5.3

As shown in Figure 9, the double-layer flexible support technique adopts two lengths of steel strands. The first layer of flexible support is used to construct the thick fundamental anchorage layer, and the second layer of flexible support is used to realize the thicker reinforced anchorage layer. The advantages of this technique include the following:

(1) fully utilizing the high load-bearing capacity of the steel strands; (2) by using the thick fundamental anchorage layer, a certain thickness of rock layers on the roof is consolidated into one unit, and anti-disturbance support is achieved; and (3) by using the thicker reinforced anchorage layer, the stability of the deeper rock masses can be fully mobilized in order to constrain the deformation of shallow, unstable rock masses, thereby further improving the anti-disturbance performance of the support structure.

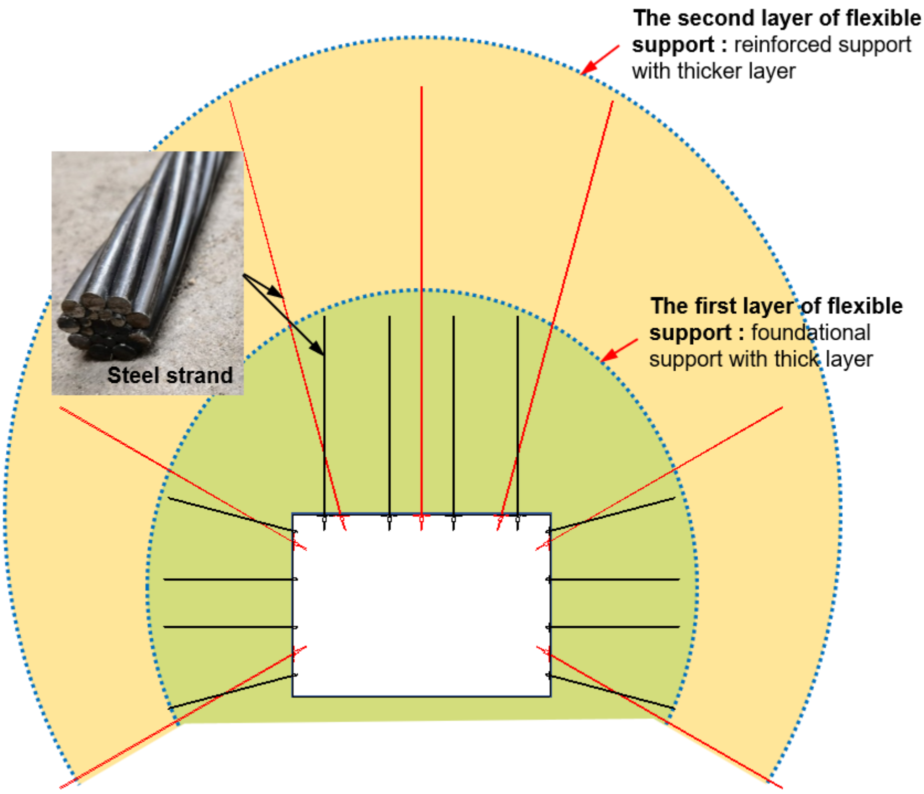


Figure 9. Supporting technique with double-layer flexible thick anchorage.

4.3. Support Thickness, Pre-Tightening Force, and Key Positions

In order to ensure that the anchored rock beam of the roadway roof is not damaged, it is necessary to ensure that σ_{max} cannot exceed its tensile strength σ_t (MPa). Based on Equation (3), the safety condition criterion for the roof can be obtained as follows:

$$h \geq \sqrt{3qw^2/4b\sigma_t} \tag{4}$$

Equation (4) can be used to determine the thickness h of the roof support. This means that, if h satisfies this inequality, the tensile stress on the roof will not exceed its tensile strength, thus ensuring the safety of the roof.

There is consensus within the engineering community that a high pre-tightening force should be applied to the bolt support of coal mine roadways to ensure the support effect [26–28]. According to the technical specifications for rock bolting in coal mine roadways [29], the pre-tightening force of the anchor bolt should reach 30% to 60% of that of the yield force, and the anchorage force of the anchor bolt should reach the standard value of the yield force. In accordance with this requirement, the pre-tightening force of anchor cables with a diameter of 21.8 mm should be 145–290 kN (the yield force of the anchor cable is 482 kN). However, this range remains considerably extensive. In our practice, the pre-tightening force of anchor cables with a diameter of 21.8 mm should be more precise. Specifically, the pre-tightening force of anchor cables within a length of 5 m should exceed 150 kN, while the pre-tightening force of anchor cables within a length of 5 m and 8 m should exceed 200 kN.

The deformation of the roadway mainly occurs near the midpoint of the four sides of the rectangular cross-section [30,31]; therefore, the corresponding support center of gravity must tilt towards this area. However, it cannot be ignored that there are stress concentration zones on the outside of each of the four corners of the rectangle. Due to structural effects, the anchorage performance of the rock mass in this area is relatively high. Thus, a part of the anchor cable anchored near the midpoint of the four sides can be anchored in the rock mass outside of the four corners.

5. Application Cases and Effect Analysis

5.1. Support Plan for Engineering Test Points

Working face 21405 of the Hulusu coal mine was taken as an engineering test site, and its basic conditions were as described earlier. According to the requirements of Equation (1), the excavation work of the withdrawal roadway should be completed at a distance of 171 m from the coal mining face. In fact, the excavation time of the withdrawal roadway was advanced, and when the excavation was completed, the working face was still 300 m away. Therefore, the withdrawal roadway was free from the disturbance of the coal mining face during the excavation period.

According to the requirements of Equation (4) and the parameters provided in Section 4.1, the thickness h of the roof support varies with the tensile strength σ_t of the roof, as shown in Table 3. The tensile strength range of the roadway roof at this test point is 1.8–2.4 MPa. In order to ensure the safety of the mining production from the most unfavorable perspective, the tensile strength of the roof should be calculated as 1.8 MPa, and the thickness of the roof support should not be less than 3.4 m.

Table 3. The thickness of the roof support varies with the tensile strength of the roof.

σ_t (MPa)	0.2	0.4	0.6	0.8	1.0	1.2	1.4	1.6	1.8	2.0	2.2	2.4	2.6	2.8	3.0
h (m)	10.2	7.2	5.9	5.1	4.6	4.2	3.8	3.6	3.4	3.2	3.1	2.9	2.8	2.7	2.6

A support plan for the withdrawal roadway was designed, as shown in Figure 10, as follows: (1) The first layer of support for the roadway roof adopted a Φ 21.8 × 4300 mm flexible steel strand support, with a pre-tension of 200 kN. The second layer of support adopted a Φ21.8 × 8300 mm flexible steel strand support, with a pre-tension of 240 kN. (2) The first layer of support for the non-mining rib of the roadway adopted a Φ 22 × 2500 mm threaded steel anchor bolt support, with a pre-tightening force of 60 kN. The second layer of support adopted a Φ21.8 × 5300 mm flexible steel strand support, with a pre-tension of 200 kN. (3) The mining rib of the roadway adopted a Φ 22 × 2500 mm threaded steel anchor bolt support, with a pre-tightening force of 60 kN. When the working face arrived, the shearer cut away the mining rib, so the flexible steel strand support was not used. (4) The threaded steel anchor bolt and flexible steel strand were both anchored with MSCK2370 resin coil. (5) Metal mesh protection was employed for the roof and roadway sides, and W-shaped steel strips were also used to strengthen the protection.

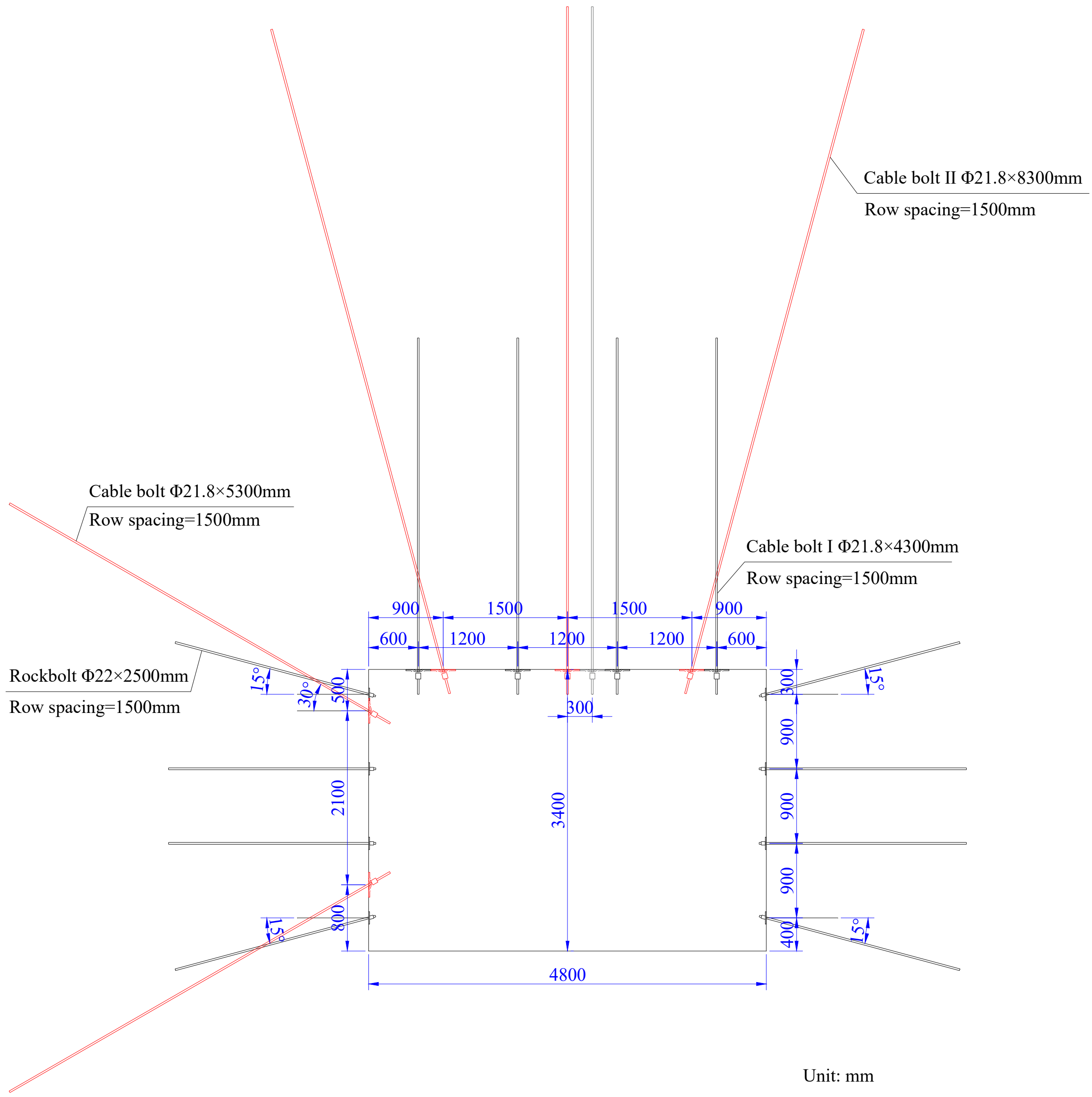
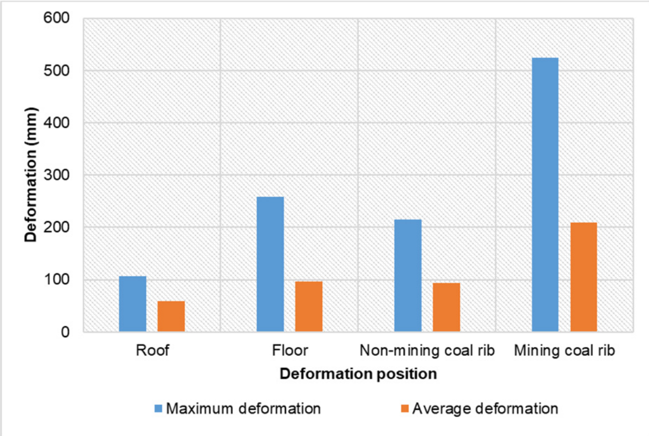


Figure 10. The support parameters of withdrawal roadway 21405.

5.2. Field Application and Effect Analysis

The active roof control technique of double-layer flexible thick anchorage fully utilizes the high load-bearing performance of deep rock masses to constrain the deformation of shallow rock masses, achieving the deformation linkage of deep and shallow rock masses. Figure 11 shows the supporting effect of the field application. The deformation of the surrounding rock was measured using the 'cross-section method' in the central area of the withdrawal roadway using steel measuring tape and laser rangefinders. The deformation includes two stages of excavation and mining. After strong mining, the maximum deformation of the roof, floor, non-mining rib, and the midpoint of the mining rib are 107 mm, 258 mm, 215 mm, and 524 mm, and their average deformation is 59 mm, 97 mm, 93 mm, and 209 mm, respectively. The cross-sectional dimensions of the roadway after deformation fully comply with the usage requirements and exhibit a relatively ideal deformation control effect.



(**a**)

(**b**)

Figure 11. The effect of field application in withdrawal roadway 21405: (**a**) The deformation; (**b**) A photograph of withdrawal roadway 21405 after mining.

This technique effectively solves the problem of controlling large deformation in roadways under strong mining disturbance, greatly supporting and effectively ensuring the continuity of mine production. Moreover, this technique simplifies the support process, eliminating the heavy workload and safety risks associated with traditional stacked support transportation. As a result, it offers significant economic and technical advantages compared to the original technology.

6. Conclusions

(1) The occurrence mechanism of the large deformation of the surrounding rock in a withdrawal roadway during mining activities has been revealed, which includes three aspects of the influence of front abutment pressure on the coal mining face, including the stress superposition exceeding the bearing capacity of the surrounding rock, the damage accumulation of the surrounding rock in the withdrawal roadway during a long mining cycle, and the fluctuation evolution of mining-induced stress. Among them, the fluctuation evolution of mining-induced stress has often been overlooked in the past, but it may be the most important cause of the large deformation and instability of the roadway surrounding rock.

(2) A method has been proposed to determine the excavation timing of the withdrawal roadway. The adoption of the proposed method not only allows for a reasonable project arrangement during the construction period, but also avoids the impact of advanced mining of the working face on the withdrawal roadway during excavation. From the perspective of facilitating the maintenance and control of the withdrawal roadway, the sum of the influence ranges of the abutment pressure between the working face and the roadway can be selected as the safe distance to avoid significant disturbances.

(3) The anti-disturbance principle of thick-layer anchorage has been clarified. The single layer of the roadway roof is characterized by its thinness, lack of stability, and susceptibility to deformation. When multiple thin rock layers are transformed into thick and well-integrated anchored layers through anchor support, the stability and disturbance resistance of the roof can be greatly improved. The thickness of the anchored rock beam is a sensitive factor that affects the maximum subsidence and maximum tensile stress of the roof and is crucial for ensuring the safety of the roadway roof.

(4) A novel double-layer flexible support technique has been proposed. In this support scheme, 1860-MPa-grade flexible steel strands are used as the carrier, overcoming the limitation of the rigid anchor bolt length limited by the roadway height, and achieving thick anchorage. The thick fundamental anchorage layer is constructed with the first layer of flexible support, and the thicker reinforced anchorage layer is constructed with the

second layer of flexible support. Through this double-layer flexible and thick anchorage, the mining disturbance to the withdraw roadway can be effectively resisted. Specific explanations have been provided for the thickness, per-tightening force, and key support positions of the support. The calculation method of the thickness h of the roof support has also been also proposed.

(5) Based on the application site of the withdrawal roadway in working face 21405 of the Hulusu coal mine in Inner Mongolia, China, the technical effect verification has been provided. The withdrawal roadway has a buried depth of 650 m, a coal thickness of 3.4 m, a rectangular cross-section shape, a width of 4.8 m, and a height of 3.4 m. The application of new double-layer flexible support technology has achieved ideal results in controlling roadway deformation after the completion of mining.

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